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UNITED STATES
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BUREAU OF MINES
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MILLING METHODS AND COSTS OF THE CARDINAL GOLD MINING CO., BISHOP CREEK, CALIF.

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WALTER B. LENHART

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UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

MILLING METHODS AND COSTS OF THE CARDINAL GOLD MINING CO., BISHOP CREEK, CALIF.[1]

By Walter B. Lenhart[2]

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2/ One of the consulting engineers, Bureau of Mines, and mill superintendent, Cardinal Gold Mining Co.

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INTRODUCTION

This paper is one of a series on milling methods and costs being published by the Bureau of Mines. The milling practices and special problems relating to grinding, tailings disposal, and choice of a treatment process at the Cardinal mill are discussed.

The property of the Cardinal Gold Mining Co. is about 18 miles southwest of the town of Bishop, Inyo County, Calif., at an altitude of approximately 8,500 feet. The camp has been designated Bishop Creek by the postal authorities. The property comprises 33 mining claims, including fractions and millsite in sections 19, 20, 29, and 30, T. 8 S., R. 31 E. These claims are located in a general way along the strike of a band of quartzite.

ACKNOWLEDGMENTS

Permission to publish this paper was granted by Victor Bongard, general manager of the Cardinal Gold Mining Co.

HISTORY

Little historical information on the Cardinal mine prior to 1908 is available, but during that year a few diamond-drill holes were drilled.

A small cyanide and amalgamation plant (now dismantled) was erected in 1915; about 5,000 tons of ore treated was said to have yielded $11.40 per ton (gold at $20.67 per ounce). Amalgamation was unsuccessful and the plates were replaced by tables. About 50 percent of the gold was recovered by the tables and 35 percent by cyanidation of the tails. The plant capacity was 75 tons per day.

From 1915 to 1919 the property was idle, but in 1919 the flow sheet was changed to all flotation and was operated until 1922. About 27,000 tons had been treated by the end of this period. It is said that the ratio of concentration was about 25 to 1 and recovery 90 percent. Concentrates were shipped to Midvale, Utah. Ore treated contained $8 to $12 per ton (gold at $20.67 per ounce).

During this period the ore came from shallow depth and was partly oxidized material. It contained a smaller amount of sulphide minerals than the present ore.

In October 1933, the property was optioned to the Consolidated Metals Corporation, which, during the same year, turned the property over to the Cardinal Gold Mining Co. During the fall of 1933, the Cardinal Gold Mining Co. started the erection of a 150-ton per day all-flotation mill. The mill was scheduled to start February 15, 1934, but on February 11 caught fire and was destroyed. A contract was immediately let to erect a new 150-ton capacity mill of fireproof construction. When completed, this mill was

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placed in operation; the first shipments of concentrates were made to the Midvale plant of the United States Smelting, Refining & Mining Co. in August 1934. About 18 months after operations began, a second grinding unit was added to the mill, bringing the milling capacity up to 300 tons per day, and 6,000 tons of ore are now being treated monthly. The mine (September 1937) had reached a vertical depth of 600 feet.

GEOGRAPHY AND ACCESSIBILITY

The nearest railroad is at Laws, Calif., 5 miles northeast of Bishop, at an altitude of 4,115 feet. The railroad is a narrow-gage, single-track line owned by the Southern Pacific; it connects Owens Valley with the Southern Pacific standard-gage road at Mina, Nev. Telephone lines and paved road connect the mine with Bishop. The road also serves four Southern Sierra Power Co. hydroelectric power plants spaced along Bishop Creek below the mine; the road is kept open the year round by the California State Highway Department.

TOPOGRAPHY

The topography of the region is extremely rugged, with high mountains deeply scarred by recent glaciation. Bishop Creek Canyon, in which the mine and mill are situated, shows a characteristic U-shaped profile; steep walls capped by serrated ridges rise on each side to thousands of feet; these prominences are snow-covered most of the year. Hanging valleys, tarns, and remnants of glacial moraines are common.

CLIMATE

The lower altitudes are essentially semidesert, as rainfall is moderate. In the higher regions melting snow supplies ample water for domestic, milling, and hydroelectric purposes. The maximum temperature during the summer is about 85° F.; winter temperature as low as 22° F. below zero has been recorded. During the winter the amount of snowfall is considerable; the deepest single snowfall recorded was 66 inches. Usually about 5 feet of packed snow covers the area around Bishop Creek during the cold season.

GEOLOGY

The prominences in the vicinity of the Cardinal mine are chiefly granite and monzonite of the Sierra Nevada granodiorite batholith. These rocks enclose a roof pendent of quartzite striking northwest and southeast and extending deep into the batholith. The dip is 70° northeast to almost vertical. The belt of quartzite cuts across the canyon of Bishop Creek and can be traced for several miles in each direction in an uninterrupted line. The quartzite band is 400 to 6,000 feet wide and is reddish-brown, which causes it to stand out in distinct contrast to the gray granitic rocks.

The ore shoots occur entirely in the quartzite and contain sulphides, wollastonite, and reddish garnets in abundance. Silicification and

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chloritization are noticeable. Mineralization, in the order of abundance, consists essentially of pyrrhotite, arsenopyrite, pyrite, chalcopyrite, and sphalerite. There is a small amount of free gold, but it is seen rarely in hand specimens. The gold appears to be closely associated with arsenopyrite and wollastonite, although the pyrrhotite and pyrite contain small amounts. There is a trace of silver in the ore. Walls of the ore body are not well-defined; economic limits are determined by sampling and assaying.

The ore is hard, semitough quartzite; it is more difficult to fracture than most of the quartzy Mother Lode ores.

SURFACE PLANTS

The surface layout comprises a mill, a machine-shop and compressor building, drillsteel shop, hoisthouse, crusher building, oil-storage building, an assay office, and administrative offices.

The mill, including the crushing plant, is of steel construction, covered with corrugated iron. The interiors of the buildings are lined with gypsum wallboard; during the coldest weather it is quite comfortable within. The bins are wood.

The compressor and machine-shop building houses two class WN-3, Sullivan, angle, compound compressors, each having a capacity of 1,340 cubic feet per minute, that deliver air to the mine. One of the compressors is driven by a "V" belt drive and the other by an endless leather belt. The machine-shop section consists of a 20- and a 15-inch lathe, shaper, power hacksaw, drill presses, and other smaller equipment. It is a very complete shop.

In the assay office are two type 1111, Denver fireclay muffle furnaces, the usual crushing and grinding equipment and core splitter, filters, and electric driers. On an average, about 15 mill samples and 30 mine samples are assayed each day by one assayer and helper. The maximum number of samples run per day is about 90. Gold is the only metal determined. The assay office is also equipped with materials for making practically all of the ordinary wet, nonmetallic, and metallic determinations.

MILLING OPERATIONS

Figure 1 shows the flow sheet of the Cardinal mill. The mill has two grinding sections and a capacity of 300 tons daily when both sections operate. Treatment is chiefly flotation, although a high-grade concentrate is obtained from tables installed in the grinding-classifying circuit.

Crushing

Ore is hoisted by a 2-drum, No. 60 Vulcan hoist through a 2-compartment vertical shaft in skips holding 1-1/3 tons each and dumping into two wooden

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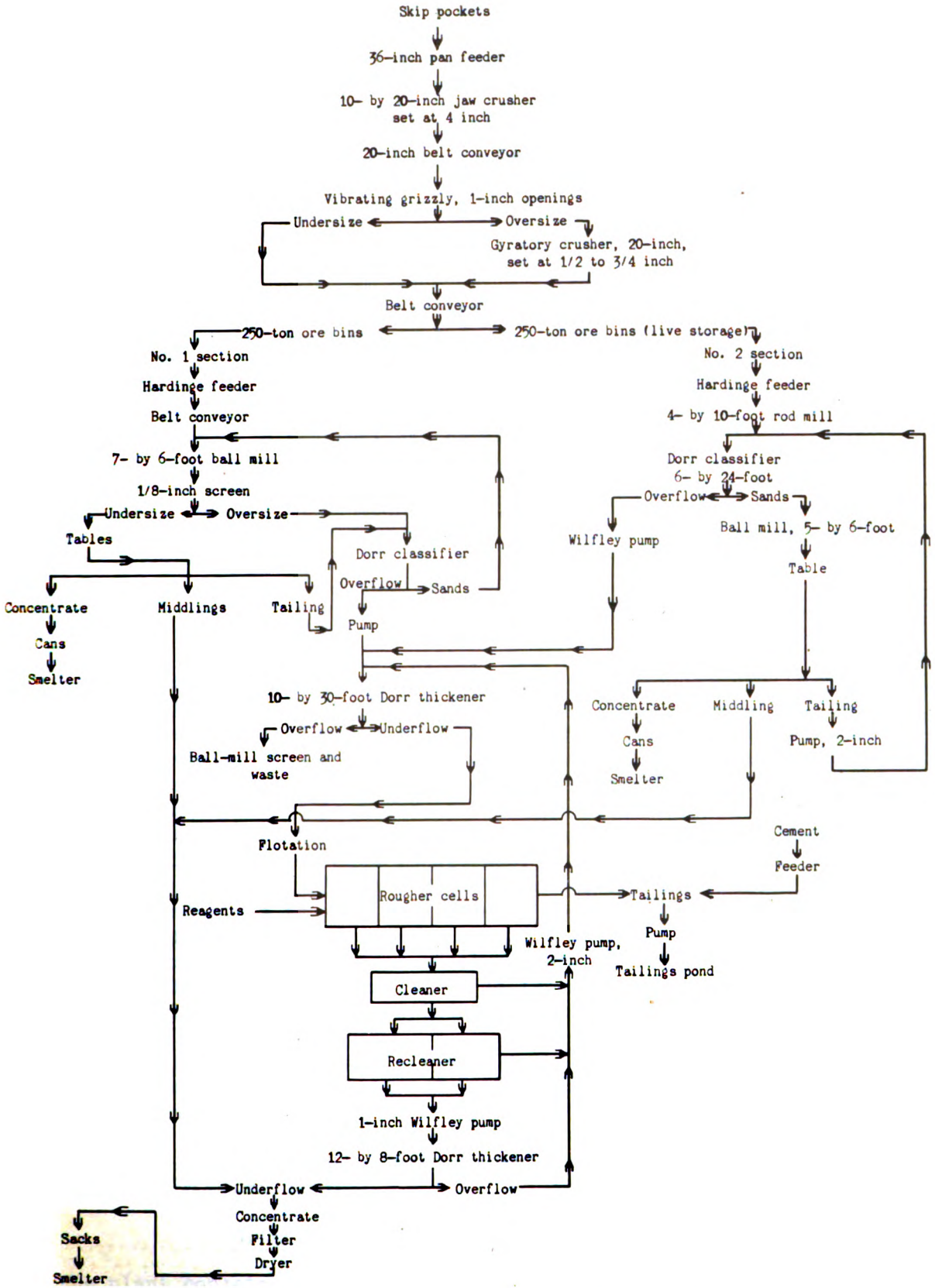

Figure 1.- Flow sheet, Cardinal mill, capacity 300 tons per 24 hours.

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bins each holding about 15 tons. The bins discharge directly into a 36-inch pan conveyor that delivers the ore to a 10- by 20-inch Allis-Chalmers jaw crusher. The crusher operator at the pay conveyor picks wood and other debris from the ore. Waste can be diverted by a butterfly gate to a third small bin alongside the other two; the waste is hand-trammed by the crusher operator to the dump. As storage ahead of the crusher is small, an operator is on shift during the two shifts that the mine operates. The butterfly gate is operated by an air cylinder and is under the control of the hoist operator. The track on the waste dump is protected from snow by a long shed for winter operations.

The jaw crusher is set to 4 inches and discharges to a short, 20-inch, inclined belt-conveyor that serves a 2- by 8-foot Jeffrey-Traylor, type 1, serial no. 1204, 110-volt, 60-cycle, electrically vibrated grizzly with bars spaced 1 inch apart. A pancake-type magnet removes tramp iron from the inclined belt to protect the gyratory crusher. The oversize from the vibrating grizzly falls to a Traylor, TY, 20-inch, gyratory crusher set to deliver a 1/2- to 3/4-inch product. The fines from the vibrating-grizzly and the gyratory-crusher product join and fall to a 20-inch, inclined belt-conveyor that operates on an 18° incline. The belt is 220 feet, center to center, and delivers the crusher product to either of two bins of laminated wood construction.

Corrugated manganese-steel jaws are used in the primary breaker. The angle of nip is kept low, but often hard boulders fly out of the jaws. Peening of the jaws is more destructive than the wear. Often a set of jaws has peened until it is too wide for the frame before it is worn out. So far, attempts to rehabilitate the crusher jaws with hard surfacing metal have proved uneconomical. However, the gyratory mantel, when worn, is economically rebuilt with a hard surfacing alloy purchased locally and called "500 hard", a trade name. Mantels are allowed to wear until the crusher discharge product is close to 1-1/2 inches, when the spindle is resurfaced with the electric arc-welding equipment. It requires about 75 pounds of welding rods for each repair job, and the rods cost 52 cents per pound. The repaired mantel lasts 4 to 5 months, crushing 25,000 to 30,000 tons of ore, and gives as good service as a new part. It requires about five 8-hour shifts to resurface a spindle. A spare is always kept on hand. The concaves are chrome-manganese steel and are not resurfaced. Several layers of metal are sometimes required to repair a mantel. A template is used by the welder to build the spindle to its original shape and thickness.

The two ore bins are each 17 feet 4 inches by 17 feet 4 inches and 18 feet deep and each has a capacity of 5,450 cubic feet; they are flat-bottomed. Using 17 cubic feet of ore per ton, the bins hold 320 tons each, or 18 tons per foot of depth. The feed conveyor discharges at such an elevation that the bins can be coned to complete capacity, giving a live load of about 250 tons each.

Grinding and Classifying Circuit

The plant contains two separate grinding units, which can be operated jointly or singly. The older unit, referred to as no. 2, consists of a

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4- by 10-foot Stearns-Rodgers rod mill that discharges to a 6- by 24-foot Dorr classifier. The sand from the classifier passes to a 5- by 6-foot Stearns-Rodgers ball mill with a trunnion discharge. The reground sand passes over a table, the tailing from which is returned to the classifier.

Unit no. 1 consists of a 7- by 6-foot Allis-Chalmers grid-type ball mill operating in closed circuit with a 6- by 23-foot, 6-inch type "D", model "DSD", heavy-duty Dorr classifier. On the discharge trunnion of the mill is a small rotary screen 15 inches in diameter by 24 inches long of 1/8-inch square-mesh wire cloth. A portion of the overflow water from the large Dorr thickener is sprayed onto this screen; the undersize passes to an 11-D Wilfley concentrating table. The tailing from the table, together with the screen oversize, passes to the classifier. Each grinding unit is served by a Hardinge constant-weight feeder in conjunction with a short conveyor belt. This makes a very satisfactory arrangement.

The rod mill is loaded with 6 tons of 4-inch-diameter steel rods. Each rod weighs 330 pounds. The larger ball mill is loaded with 11 to 12 tons of 4-inch pressed-steel balls and the smaller mill carries 4 to 5 tons of 4-inch balls. Liners in the rod mill and both ball mills are manganese steel; they last about 7 months of continuous operation. Liner consumption is about 4 pounds per ton of ore. The larger grid-type ball mill consumes 3.83 pounds of balls per ton of ore ground; the rod mill, 1.92 pounds per ton; and the smaller ball mill, 1.31 pounds per ton. Since the total rod and ball consumption in no. 2 unit is 3.23 pounds per ton of ore ground, it appears that the rod mill-ball mill set-up is the more efficient grinding unit.

Balls are added each day, the amount depending on the tons crushed. Three and seventy-five hundredths pounds per ton ground customarily are added to the grid-type mill; 2 pounds per ton ground are added to the smalle mill. Rods are added at the rate of one per day, two rods being added on alternate days. At least every 30 days the ball mills are opened for inspec tion of ball load and liner wear.

By using 4-inch balls in conjunction with manganese-steel liners, it is observed that better wear is obtained from the liners by the use of the larger balls, probably due to the greater peening and surface working quality of manganese steel. This property of manganese steel is discussed by Williams and Homerberg3/.

Liner bolts for all the mills are fabricated in the mill shop. Thread- ing is accomplished by a special chuck that operates in a large drill press A satisfactory gasket assembly is also made locally. A ring about 3/16-inch high is first cut from heavy 2-1/2-inch pipe. Three 1/8-inch rubber gaskets are then cut to fit over the liner bolt and inside the metal ring. Between the nut and the rubber gasket is imposed an ordinary metal washer. Great pressure can be exerted on the liner bolts, and the gaskets will not feather out.

3/ Williams, Robert S., and Homerberg, Victor O., Principles of Metallography: McGraw-Hill Book Co., Inc., 1935, pp. 144-146.

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The overflows from both classifiers are pumped to a 30- by 10-foot Dorr thickener. Each unit can grind 160 to 170 dry tons of ore per 24 hours, depending on the relative fineness of the ore, ball load, and other minor factors.

No. 1 classifier overflow is pumped to the thickener by a 3-inch, rubber-lined, Hydroseal pump; the sand is returned to its ball mill by a screw conveyor. No. 2 grinding unit uses a 2-inch Wilfley to pump the classifier overflow to the thickener. Overflow from the classifiers has an average solid content of 18 to 20 percent.

The fineness of the products ground in the two mills is very similar. An average screen test of the classifier overflow for a 30-day period gave the following:

Mesh	Percent
Plus 100.....	4.5
100 to 150.....	14.2
150 to 200.....	20.6
Minus 200.....	60.7

Assays of the plus 200-mesh and minus 200-mesh material show that the gold loss in the tailings is about equally divided between the two sizes, when the respective weight of each product is considered.

About once a month the classifiers are thoroughly cleaned and the sands passed over the tables. Each classifier will produce about 20 ounces of gold in a high-grade concentrate, which is shipped in cans along with the regular high-grade shipments. Cleaning out the classifiers prevents free gold from accumulating in the grinding circuit and removes from the classifier considerable tramp iron, chiefly small balls and bits of broken rod that eventually would impede the mechanical efficiency of the units. It has been noticed that the greatest accumulation of free gold is not at the extreme bottom or toe of the classifier, but at the water line where the rakes emerge from the pulp; the concentration here is probably due to the wave action of the pulp.

The table produces a high-grade concentrate that is shipped in sealed cans and a middling maintained at about 10 ounces of gold per ton which is shipped with the filtered concentrates. About 40 percent of the recovered gold is obtained on the table from its two products.

Table concentrates are shown by analyses to be mostly arsenopyrite. Some pyrite that shows on the table is cut with the middling, along with some arsenopyrite. The middlings contain a small amount of garnet and little or no pyrrhotite.

The purpose of the tables is to prevent any excess of free gold from building up in the grinding circuit. The tables have the disadvantage of

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robbing the filtered concentrate of some of its metal, thereby lowering its grade. When the mill heads are below 0.25 ounce gold to the ton, it is customary not to run the tables, as the practice lowers the grade of the flotation concentrate too drastically. From a strictly metallurgical standpoint, tables do not affect the ultimate extraction, except that more water gets into the system and helps to overload the Dorr thickener.

Thickening

The underflow from the 10- by 30-foot Dorr thickener passing to the flotation cells is kept at 40 to 45 percent solids. The overflow from the thickener contains a small amount of solids, but the gold loss from this source is considered small.

The thickener has an area of 700.69 square feet. When grinding is done with one unit there are 4.5 square feet per ton per 24 hours; at 300 tons per day there are 2.25 square feet per ton per 24 hours, an area not quite enough to maintain a clear overflow.

The overflow with one mill is about 30 gallons per minute; with both mills it is 60 gallons per minute. As some of the overflow is used for make-up water in the rod and ball mills, the average flow will be considerably less than this figure.

The use of the Dorr thickener ahead of the flotation cells gives some storage capacity, because the thickener acts as a surge tank. With all the grinding units down, the cells can continue to operate for a short time. This practice is inefficient, as the percentage of solids in the pulp diminishes and the gold content of the tailings rises rapidly.

Several times when grinding was disrupted samples of the tailings were taken and their solid and gold content determined. The curve shown in figure 2 illustrates approximately what can be expected when the solids in the flotation pulp are allowed to drop.

Flotation and Filtering

The thickened pulp flows by gravity to the head end of a bank of Fagergren flotation cells. The first cell in the series, a rougher, is a 56-inch-square unit equipped with skimmers. The next three roughers are each 56-inch-square Utah-type cells without skimmers. The rougher froth flows to a square 45-inch cell for cleaning, and the cleaner froth is, in turn, recleaned in two 36-inch Utah-type cells.

The middlings from the cells can be pumped either to the classifiers or to the 10- by 30-foot Dorr thickener, but as most of this material is minus 200-mesh, it is usually pumped to the thickener. There is a 2-inch Wilfley pump for either purpose.

The recleaned froth flows to a sump, where it is pumped by a 1-inch Wilfley to a 12- by 8-foot Dorr thickener. The thickened concentrate pulp

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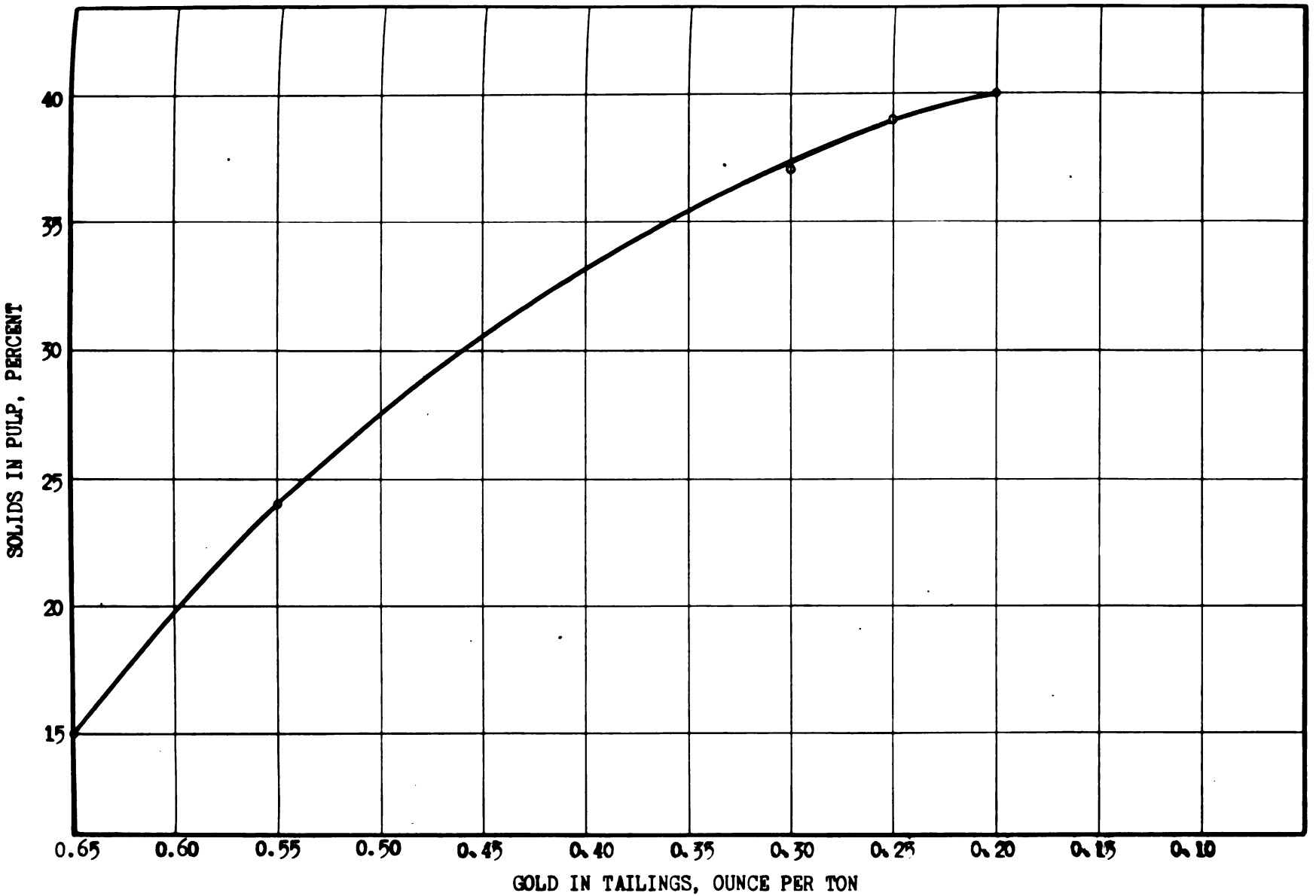

Figure 2.– Effect of dilution on residual gold in tailings.

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is lifted by a Dorrco diaphragm pump and sent to a two-leaf, 4-foot American filter, which has an effective filtering area of 44 square feet. The vacuum receiver is discharged by a small centrifugal pump.

A novel method is used to produce the vacuum at the filter. Most metallurgists are familiar with the ordinary laboratory suction filter that is affixed to the hydrant. At the Cardinal, this same principle is used for obtaining the vacuum. The device has been in continuous use for about 8 months and has filtered all the concentrates produced. Repairs have been unnecessary.

The cell tailings flow to a sump, where they receive a predetermined amount of portland cement, for settling purposes, and are pumped to a tailing pond by a 3-inch rubber-lined pump.

Gold appears to be associated chiefly with arsenopyrite; when that mineral is found in the tailings by panning, the gold content rises. Massive pyrrhotite assays about 0.03 ounce gold per ton and predominates in considerable excess over the total remaining sulphides; it slimes much easier than the tougher arsenopyrite. By cleaning and recleaning the rougher froths at a pulp density of 20-percent solids, most of the pyrrhotite is eliminated. Spraying the froth to wash out the pyrrhotite from the interstices of the bubbles dropped some of the arsenopyrite and proved a decided detriment.

Reagents

Up to about a year ago it was customary to use xanthate (Z-5) and 301 as conditioners, with pine oil as a frothing agent. The amounts used ranged from 0.20 to 0.30 pound each of Z-5 and 301 per ton treated, or a total of 0.4 to 0.6 pound of conditioner per ton treated. This was considered excessive. Pine oil amounting to 0.10 pound per ton, sufficient to maintain a proper froth, was used. A long series of experiments was undertaken; each week the amount of xanthate and 301 was reduced. At present, 0.03 pound of Z-5 and a similar amount of 301 are used. Pine oil has been replaced by Aerofloat No. 15, which is used at the rate of 0.10 pound per ton. The weekly reduction in the amount of conditioner reagents still continues. If anything, there has been a slight improvement in extraction and a decided reduction in the cost of reagents.

Under the old schedule it was practice to mix 30 pounds of Z-5 and 30 pounds of 301 in 40 gallons of water. This strength of solution was found to cause some segregation. At present, 5 pounds of Z-5 and 5 pounds of 301 are mixed with 50 gallons of water. When one unit is run, this will last about 24 hours.

The changes in reagents and amounts have not affected the grade of concentrate, although the percentage of insoluble is now higher than formerly.

The use of sodium silicate, sodium sulphide, copper sulphate, copper sulphate derivatives, soda ash, and other reagents gave no harmful or bene-

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ficial results. Feeding the different reagents in the various places in the grinding and classifier circuits has not been beneficial. By feeding the reagents at the head of the cells, frothing in the grinding and thickening circuits is eliminated.

All of the reagents are fed at the head end of the cells. Aerofloat No. 15 is fed with a Clarkson-type feeder, and a Z-5, 301 mixture is fed by a drip feeder. When No. 239 is used, it is fed into the second rougher cell.

Handling Concentrates

Concentrates from the filter fall to a 3-compartment steam drier; they are sacked when the moisture content is about 10 percent. The drier, which was made in the company shops, consists of an electrically welded rectangular hopper, the lower half and bottom of which are provided with a steam jacket. The entire assembly is mounted on car wheels and can be moved in front of the filter when the compartment in use is filled.

The flotation concentrates, when dried to 10 to 12 percent moisture, oxidize in burlap sacks so rapidly that the sacks are often completely destroyed by the heat. A carload of concentrates once caught fire from this cause. To overcome this difficulty, heavy paper-lined sacks are used, but a lighter paper-lined sack can be used in the colder months. Further drying would be beneficial but, due to the drier capacity and amount of concentrates made, is uneconomical.

Steam for drying concentrates and for heating the mill is provided by a small fire-tube boiler made by the American Radiator Co. In this same room is a small Ideal 08-1M water heater for heating water for the shower baths. The condensed steam from the heating radiators and from the concentrate drier is returned to the boiler.

The concentrates are sacked by hand and trucked by the company to Laws, where they are loaded onto narrow-gage cars.

All shipments are sent to the Utah Sampling Works, which samples for the shipper and smelter. The Sampling Works provides three samples, one for each interested party and one to be retained for umpire purposes.

Disposal of Tailings

Tailings have a density of 42 to 45 percent solids. The density of the pulp flow is increased 2 to 3 percent by removal of liquid in the flotation concentrates. Tailings are pumped out of the mill by means of a 3-inch rubber-lined pump a vertical distance of about 16 feet, after which the material flows in a 3-inch pipeline to the tailing pond, a distance of about 2,000 feet. The pipeline receiving the tailings is laid on a uniform downgrade to permit drainage should the flow of liquid stop during cold weather.

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The 3-inch line is carried on a simple trestle at the pond about 35 feet back from the outer edge of the bank. At 50-foot intervals, a "T" and a 2-inch valve have been installed. The valve is connected by a hose to a 20-foot length of 2-inch pipe in which 16 to 18 holes have been drilled along its length. The pipe, locally called a "spreader", is placed at the outermost portion of the dam; as the pulp escapes from the holes, the heavier and sandy portions build up in front of the spreader. Three spreaders are in constant operation and an attendant on each shift moves them from point to point, thereby building up the outer edge to a uniform height. The attendant keeps the outer edge built up one or two feet in height by shoveling.

An "L" in the mountain side forms a side and one end of the rectangular tailing-pond area. An 8-inch steel-pipe outlet or drainage pipe runs on the ground under the pond back almost to the edge of the hill. A vertical riser made up of a series of 6-inch T's and close nipples has been installed at the upper end of the pipe. As the pond grows in height, the T's are plugged off so as to drain only the uppermost film of water.

When the dam reached a height of about 50 feet, a second drainage pipe was installed at that elevation. It is now in use and serves mostly as an emergency outlet should the lower pipe become plugged. The outer edge of the pond at its highest point is 65 to 75 feet high.

The tailing pond now in use is 700 by 200 feet at the surface; it has an area of 140,000 square feet. About 100,800 square feet is effective settling area; the rest contains the coarser sand that settles out before reaching the liquid area. As the average mill tonnage has been about 200 tons per day, the pond has 504 square feet per ton of ore milled or about 520 square feet per ton of tailings. At times as high as 340 tons per day has been milled, and the pond handles that tonnage without difficulty. This is equivalent to 297 square feet per ton milled or 315 square feet per ton of tailings.

As a settling agent, dry portland cement is added to the tailing at a rate ranging from 1 sack per 24 tons milled to 1 sack per 60 tons milled, depending on conditions at the tailing pond and on the amount of water in Bishop Creek, which receives the tailing-pond effluent. The average is about 1 sack per 40 tons milled.

The cement is fed into the tailing stream by a belt feeder that delivers a uniform amount; it has been found unnecessary to feed the cement each hour of the day. It is customary to feed the allotted amount of cement each shift over a 2- to 3-hour period, after which the feeder is shut off until the following shift. Although portland cement probably helps to stabilize the pond, it is not added for that purpose, but to promote settling and provide a perfectly clear overflow.

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Re-treatment of Tailings and Clean-up Treatment

To date, 161,316 tons of tailings have been stocked in two ponds; approximately 155,000 tons are in one pond. The larger pond will continue to be used for some time. The tailing ponds contain 4,942.0 ounces of gold, or an average of 0.0307 ounce per ton. The earlier tailings from former operations, amounting to about 13,000 tons assaying 0.09 to 0.13 ounce gold per ton, are now being re-treated in the present mill. The tonnage treated is small and will average about 900 tons per month, although this may be increased later. Tests are now under way to recover gold from the flotation tailings. So far, it appears that slime tables might make an economical recovery. Tests made to recover the gold on corduroy have been unsuccessful.

The area around and under the old mill that was built in 1915 (now dismantled) was cleaned up by the company in 1934, and gold to the net value of $5,242 was recovered. The clean-up material was sacked and shipped direct to the smelter. A second clean-up of this area is now under way, and indications point to a substantial net recovery. The material now being treated is so saturated with old reagents that refloating is impracticable. Treatment comprises disintegration and tabling.

Mill Control

A 200-pound head sample is taken from each mill and cut to 50 pounds by the mill crew for delivery to the assay office during each 24-hour period. A 24-hour sample as well as a shift sample is taken by an automatic tail sampler.

A 24-hour shift sample of the concentrates is taken before filtering; each shift takes a separate sample of the same material. These samples guide the operators in regulating the grade of concentrates.

The table concentrates and table middlings are sampled once each day while being prepared for shipment. Table middlings are then shipped in the same lot with the flotation concentrate. Two men who sack the filtered concentrates also take a sample of the dried concentrates by means of a 2-inch auger. In addition, as they sack the material they take a hand sample from every sixth sack. These are assayed each day, so that by noon the results for the previous day are available.

The assayer makes a composite sample of all shipments based upon the daily production records. This composite is assayed when the lot is closed. In addition, the daily auger and hand samples that make up the composite are assayed each day. This serves as a check on the calculated metal content of a shipment.

The high-grade concentrate is stored and shipped once a month. Before shipment this material is resampled and weighed. The material usually is shipped by express, although it has been trucked direct to the smelter at Midvale.

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The filtered concentrates are weighed and their moisture content determined each day, after which the sacked material is trucked to Laws. On account of the rapid oxidation of the pyrrhotite, it is desirable to ship the products as quickly as possible, as long standing aids in the destruction of the sacks. It is customary to ship 1 car per week.

As soon as the assay reports are available, one copy is sent to the mill, one is retained in the assay office, and a third is delivered to the office. In the office a daily calculation is made to show the following:

Mill heads: Tons, assay, ounces gold therein.
Flotation heads: Tons, assay, ounces gold therein.
Flotation tails: Tons, assay, ounces gold therein.
Flotation concentrates: Tons, assay, ounces gold therein.
Table middlings: Tons, assay, ounces gold therein.
Table concentrates: Tons, assay, ounces gold therein.
Shipments: Tons, sacks, assay, ounces gold therein.

These calculations are theoretical, being based on what should have been produced as calculated from the tonnage and assays. In addition, a second record is kept of the tons and assay values of what was actually produced.

A mill record of each lot making up a shipment is kept; this shows the tons and assay values of each daily portion of the shipment and provides the calculated weighted assay, which is checked against the composite sample prepared by the assayer.

Regular Ph determinations were made over a considerable period of time, but as they gave no useful information they are now made only on special occasions. The daily Ph determinations ranged from 7.8 to 8.2.

The flotation operator makes hourly determinations of the flotation pulp density according to the following procedure and table. The table is based on the net weight of pulp in a 1-quart milk bottle. The bottle is standardized by adding 943 grams of water and marking the top of the water. Thereafter the bottle is filled with pulp to this mark and the table applied. It will apply to any container standardized in the same manner where the specific gravity of the solids is 3.2.

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Net weight, grams	Solids, percent	Net weight, grams	Solids, percent
977	5	1,218	33
984	6	1,230	34
992	7	1,241	35
997	8	1,252	36
1,005	9	1,265	37
1,013	10	1,276	38
1,020	11	1,288	39
1,028	12	1,300	40
1,035	13	1,313	41
1,043	14	1,325	42
1,051	15	1,337	43
1,058	16	1,352	44
1,067	17	1,365	45
1,075	18	1,377	46
1,084	19	1,392	47
1,092	20	1,405	48
1,102	21	1,420	49
1,111	22	1,435	50
1,120	23	1,466	52
1,128	24	1,498	54
1,138	25	1,520	56
1,148	26	1,565	58
1,158	27	1,604	60
1,167	28	1,642	62
1,177	29	1,683	64
1,186	30	1,733	66
1,197	31	1,770	68
1,208	32	1,815	70

The gravity of the pulp is controlled at the large Dorr thickener. The diaphragm pump on the thickener was eliminated and the underflow taken directly to the cells by gravity through a 4-inch line. About a 5-foot difference in elevation exists between the top of the thickener and the outlet to the cells. At the cells the outlet is bushed down to 2 inches. A series of 8 bushings graduated in opening diameter ranging from 3/4 to 1-1/2 inches is provided. If the gravity of the pulp is too low, a smaller bushing is inserted in the discharge; this cuts down the volume of the outlet from the thickener, which automatically raises the gravity. The arrangement has proved more satisfactory than a valve, which soon cuts out from the sand.

Power

The Cardinal Gold Mining Co. had a prior water right on Bishop Creek, which was released to the Sierra Power Co. in exchange for power, so a favorable rate is obtained. The first 300 horsepower is free. The charge for the next 1,200 horsepower is $3 per horsepower-month, or 5-1/2 mils

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per kw-hr. All power over 1,500 horsepower is charged for at the regular rates. The tabulation shows the connected load and the motor horsepower for various equipment.

Mine

	Horsepower
No. 1 sinking pump	15
No. 2 sinking pump	15
500-foot level no. 3 centrifugal pump	125
Spare pump, 500-foot level	125
Triplex pump	75
Spare triplex pump	35
Sinking hoist	75
Mine hoist	75
Total	540

Surface, other than mill

	Horsepower
Compressor no. 1	200
Compressor no. 2	200
20-inch lathe	5
Shaper	1-1/2
Drill press	3
Fresh-water pump (spare)	10
Emery wheel	1
Grinding wheel	1/4
Concrete mixer	3
Cooler (boarding house)	1-1/2
Battery charger	2
Assay office	7
Timber framing saw	5
Steel shop emery wheel	5
Electric welder MG set	10
Camp use (heating and lighting of staff houses, bunk house, etc.).	1/ 100
Total	554

1/ Estimated.

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Mill

	Horsepower
Jaw crusher	20
Magnet	7-1/2
Gyratory crusher	35
Jaw crusher conveyor	3
No. 1 ball mill	150
No. 2 do.	100
Rod mill	75
No. 1 flotation cell	10
No. 2 do.	10
No. 3 do.	10
No. 4 do.	10
No. 5 do.	5
No. 6 do.	3
No. 7 do.	3
Concentrating table	3/4
No. 1 classifier	3
No. 2 do.	3
Dorr thickener (large)	5
Concentrate thickener	5
Filter assembly	10
No. 1 feed pump	10
No. 2 do.	10
No. 3 middlings pump	10
Tailing pump (spare)	10
Tailing pump	7.5
No. 1 classifier overflow pump	10
Dryer return water pump	3
Hardinge feeders (2)	1/2
Belt feeder No. 1	1
Do. No. 2	1/3
Spare belt feeder to No. 2 ball mill	3
Main ore belt	2
Pan feeder	3
Filtrate pump at filter	1-1/2
Total	540
Grand total	1,634

All mill units, except a part of the vacuum filter assembly, are powered by individual electric motors and are either direct connected or driven through "V" belts.

Labor and Personnel

One operator and a helper on each shift look after the entire milling operation. The day-shift helper assists in sacking the concentrates; the afternoon helper washes the mill floor, brings in the 24-hour supply of

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portland cement, prepares the reagents, cleans sample buckets, and attends to other details. The graveyard-shift helper does all the greasing and oiling, except of the electrical equipment, which is done by an electrician's helper. One man is employed regularly to sack concentrates and assist in handling table products. A total of 10 men, including the 3 tailing-pond attendants, are under the direction of the mill superintendent. The day operator acts as foreman.

All routine repairs are made by the operating crew. These repairs include changing leaves on the filter, tightening liner bolts, changing pump runners and repairing pumps, and repairing flotation equipment. Other repairs are made by the repair gang that serves the mine and surface workings under the direction of the master mechanic.

The surface crew comprises about 12 men. Each day the work done by any of this crew is charged to the mill, camp, or mine. The crew is as follows:

Electrician	1
Electrician's helper	1
Welder	1
Carpenter	1
Repair man	1
Truck driver	1
Caterpillar operator	1
Bull gang	3
Machine shop	2
Total	12

In addition, there are four men in the steel shop sharpening and repairing mine steel, a drill "doctor", and a compressor operator on one shift. The drill "doctor" and one of the machine-shop men look after the compressors the other two shifts.

Following is the executive and supervising personnel of the Cardinal Gold Mining Co. in 1937:

A. J. Inderrieden, president, El Paso, Tex.
Richard H. Travers, vice-president, Los Angeles, Calif.
G. Harold Janeway, secretary & treasurer, do.
Victor Bongard, general manager, Bishop Creek, Calif.
Walter B. Lenhart, mill superintendent, do.
A. B. Weston, mine superintendent, do.
Charles Green, master mechanic, do.
J. C. B. Amos, purchasing agent and auditor, do.
Charles F. Johnson, mining engineer, do.
J. D. Jones, assayer, do.

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Metallurgical Data

The following table contains metallurgical data from January 1, 1937, to September 30, 1937:

Tons milled		43,133
Heads:		
Gold	ounce per ton	0.27
Silver and copper		Not determined.
Tailings, gold	ounce per ton	0.0291 to 0.03
Recovery of gold, flotation, percent		50
Tables,	do.	40
Total,	do.	90

Recovery of silver and copper amounts to about $400 net per month.

Ratio of concentration:	
High-grade ore	24 to 1
Low-grade ore	36 to 1
Average	32 to 1
Net water consumption	300 gallons per ton of ore.

Following is a typical analysis of a shipment of flotation concentrate:

Gold	ounces per ton	6.90
Silver	do.	1.69
Copper	percent	1.85
Insol	do.	12.8
Iron	do.	36.3
Zinc	do.	.5
Sulphur	do.	24.2
CaO	do.	1.1

A typical shipment of table concentrates is as follows:

Gold	ounces per ton	115.15
Silver	do.	22.54
Copper	percent	.30
Insol	do.	4.0
Iron	do.	38.3
Zinc	do.	.4
Sulphur	do.	16.2

COSTS

Cost of Milling

From January 1, 1937, to the end of September 1937, 43,133 tons were milled at a total cost of $60,446.80, or $1.40 per ton. The total cost includes crushing, secondary crushing, grinding, classification, flotation, handling of concentrates in the mill, tailing disposal, supervision, assay-

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ing, repair labor, supplies, maintenance, insurance and industrial accident insurance. Cost of milling for the month of June 1937 is as follows:

		Per ton milled
Tons milled	5,572	
Supervision	$ 240.00	$0.04
Labor	1,810.87	.33
Power	1,584.08	.28
Supplies	1,179.73	.21
Industrial accident insurance	81.62	.02
Maintenance insurance	69.95	.01
Assaying	110.66	.02
Maintenance	1,821.62	.33
Total	6,898.53	1.24

Cost of Pump Repairs

Three 2-inch pumps of the conventional all-metal centrifugal type and two rubber-lined pumps are used in the mill. All these handle practically the same type and tonnage of pulp. The records for the past year show that for maintaining one all-metal pump the cost for repair parts was $1 per day of operation. Costs for repair parts for the two rubber-lined pumps during the same period were less than 5 cents per day. Most of this repair cost to the rubber-lined pumps was confined to one pump that received a chunk of foreign material that damaged the runner. Barring such accidents, the repair cost per day for the rubber-lined pumps is negligible. The pump used on the tailings has been in continuous operation for over 11 months and recent inspection of the runner and liner showed practically no wear.

Trucking and Liquidation Costs

Trucking concentrates to Laws costs $2.50 per ton for the 23-mile haul. This allows no credit for the backhaul of miscellaneous supplies. A load is hauled each day. The narrow-gage cars are unloaded at Mina, Nev., into standard-gage cars and shipped to Midvale, Utah. The freight, due to transfer and mileage, is high, ranging from $10.70 to $12.50 per ton, depending on the grade of concentrate.

The liquidation charges on the concentrates for a given period were $1.3711 per ton of ore milled. These charges include sack, sacking labor, drying, trucking, freight, sampling and smelting charges.

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The costs of these items are given in the following tabulation:

Sacks	$0.0200
Sacking labor	.0400
Trucking	.0525
Drying	.0350
Freight	.4375
Sampling	.0648
Smelting charges	.2313
Smelter gold deductions	.4900
Total	1.3711

SPECIAL PROBLEMS

Grinding-Circuit Practice

Tests and calculations have been made to determine the metallurgical and economic affect of removing a high-grade gold concentrate from the grinding circuit by means of tables, jigs, or a unit cell. The relative amounts of gold associated with the various gold-bearing sulphides and the physical properties of the minerals affect the metallurgy. The net return from sales of concentrate and the relative costs for removing the high-grade concentrate or letting it go into the classifiers for later recovery also must be considered.

Metallurgical tests were carried on for 30 days on the grinding-circuit products in No. 2 section. Samples were taken of the primary rod-mill discharge, the classifier sands, and the reground product from the secondary ball mill. Samples were dried, and a composite of the screen size was assayed for gold. The following tables give the results of these tests.

Rod-mill discharge product

Mesh	Percent	Gold per ton, ounce	Gold in 1 ton of this material, ounce	Percent of total metal in size product (weighted)	
Plus 28	4.9	0.26	0.01274	5.0	
Minus 28 plus 48	16.6	.22	.03652	14.3	
Minus 48 plus 65	13.3	.20	.02660	10.3	
Minus 65 plus 100	10.4	.19	.01976	7.7	
Minus 100 plus 150	12.9	.24	.03086	12.0	
Minus 150 plus 200	12.9	.25	.03225	12.7	62.7
Minus 200	29.0	.34	.09860	38.0	

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Classifier-return sand

Mesh	Percent	Gold per ton, ounce	Gold in 1 ton of this material, ounce	Percent of total metal in size product (weighted)	
Plus 28	3.9	0.34	0.01326	2.72	
Minus 28 plus 48	17.7	.24	.04248	8.75	
Minus 48 plus 65	19.9	.23	.04577	9.43	
Minus 65 plus 100	19.1	.31	.05921	12.30	
Minus 100 plus 150	18.3	.38	.06954	14.30	
Minus 150 plus 200	10.4	.85	.08840	18.30	66.8
Minus 200	10.7	1.54	.16474	34.20	

Ball-mill regrind

Mesh	Percent	Gold per ton, ounce	Gold in 1 ton of this material, ounce	Percent of total metal in size product (weighted)	
Plus 28	0.2	0.22	0.00044	0.1	
Minus 28 plus 48	7.5	.34	.02550	4.6	
Minus 48 plus 65	12.4	.32	.05248	9.5	
Minus 65 plus 100	15.5	.32	.04960	9.0	
Minus 100 plus 150	19.0	.41	.07790	14.2	
Minus 150 plus 200	14.7	.64	.09408	17.2	76.8
Minus 200	26.7	.93	.24831	45.4	

Referring to the screen tests for the rod-mill product, it is apparent that at 48-mesh there starts a segregation of gold in the finer sizes, due to the greater friability of the mineral sulphides. At about 100-mesh, the point is reached where this preferential crushing of mineral over gangue ceases.

If the mineral is all free from gangue at minus 100-mesh, then 62.7 percent of the total values are liberated in the rod mill and would be available for jigging, unit cell flotation, or tabling.

High gold content in the circulating load in the classifier, especially in the finer sizes, indicates that before the metal can get out of the classifier it has to be ground finer than minus 200-mesh. The table, as used in the present flowsheet, apparently does not remove this fine material sufficiently to prevent it building up to 1.54 ounces gold per ton, which, in view of 0.30 ounce of gold per ton heads, is high.

The conclusion reached by considering these tests indicates that metallurgically either tables, jigs, or a unit cell following either the rod mill or ball mill would be beneficial; but when the economics are considered, there appears to be no advantage.

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Observations at the mill have shown that when the heads are higher than 0.30 ounce gold per ton, the tailings may be expected to go higher than 0.0 ounce per ton. Also, the lower the grade of concentrates, the lower the tai losses. The present use of tables has not affected the grade of tailing. When the flotation concentrates are maintained at from 4 to 8 ounces gold pe ton, there is not much difference in the grade of tailings, but when the con centrates pass 9 ounces per ton, the tailings rise rapidly. Usually, a 9-ounce-per-ton concentrate results in a 0.035-ounce-per-ton tailing, and a 10 to 12-ounce-per-ton concentrate will bring the tails still higher. When the concentrates go below 6 ounces per ton, the tonnage of concentrates produced also rises rapidly. When all the variables are considered and calculated to a conclusion, it has been determined that a 6- to 8-ounce flotation concentr is the most economical to ship.

The conclusion drawn after considering the metallurgical and economic factor involved is that the use of jigs or unit cells in closed circuits wit the grinding units would not be economical, mainly because at the Cardinal they would not make a lower tailing and any further removal of the valuable minerals from these circuits would reduce the grade of flotation concentrate below 6 to 8 ounces per ton, and the net cost of the liquidation of the concentrate would increase over the present cost.

Cyanidation of Concentrates

Ore and flotation concentrates at the Cardinal mine are not easily amenable to cyanidation, due to the continuous and rapid oxidation of the co tained pyrrhotite. Tests indicate that the arsenic content is not a serious detriment to cyanidation. By roasting flotation concentrates slowly, espe-cially at the beginning of the operation, the pyrrhotite and other iron sulphides can be oxidized, producing a clinker that cyanides quite readily with a low consumption of cyanide and lime. However, cyanidation, either of the crude ore or of the concentrates, cannot be carried on at the mine, for tailing effluents would drain into Bishop Creek. Since this water is used f domestic purposes by the town of Bishop and by the city of Los Angeles, re-gardless of whether or not contamination can be avoided, cyanidation at the mine is not advisable. Even talk of cyanidation is looked upon with suspicion. It is possible that concentrates could be trucked about 15 miles to a point where they could be treated without any danger to domestic water supplies.

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